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MINING AND MILLING METHODS AND COSTS AT THE YELLOW ASTER MINE, RANDSBURG, CALIF.

BY

CORWIN L. COOPER

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MINING AND MILLING METHODS AND COSTS AT THE YELLOW ASTER MINE, RANDSBURG, CALIF.[1]

By Corwin L. Cooper[2]

INTRODUCTION

This paper, describing the milling and cyanidation methods and costs at the Yellow Aster gold mine, is one of a series of papers being prepared by the Bureau of Mines on gold-milling practices in the United States.

Acknowledgments

Acknowledgment is made to the executives of the Anglo American Mining Corporation, Ltd., for permission to publish this article, and to A. W. Frolli, manager, and other members of the company staff, for assistance in preparing the paper.

Location

The Yellow Aster mine is located in the Rand Mountains just south of Randsburg, Calif., a small mining town near the eastern edge of Kern County; the district is in the Mojave Desert. The mine and district are served by hard-surfaced roads from the north and south; Los Angeles is 150 miles southwest. A branch of the Southern Pacific Railroad passes within 5 miles of Randsburg. Hydroelectric power is supplied by the Southern Sierra Power Co. through a substation at Randsburg, drawing current from its main 88,000-volt transmission lines which pass within 1 mile of the camp. Water for mine and mill use is pumped from wells at Coler through 7 miles of pipe.

History

The Yellow Aster mine has a long and notable history. It was discovered in 1895. The first mill - a unit of 30 stamps - was put into commission in 1898, and with the exception of 3 years beginning in 1918 when the mine was

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2/ Superintendent of the Yellow Aster Cyanide plant, Randsburg, Calif., and one of the consulting engineers, U.S. Bureau of Mines.

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shut down, milling has been carried on practically continuously. In 1900 a new mill with 100 stamps was built on the opposite side of the gulch from the original plant, which was abandoned in 1916.

Until the Anglo American Mining Corporation, Ltd., took over control of the mine in 1933, amalgamation was used exclusively for recovery of the gold in the ore. Amalgamation, both inside and outside was practiced, mercury being fed to the batteries as well as to the plates. Batteries were equipped with 35- or 40-mesh screens; the tailings were deposited on the hillside below the mill.

The records show that, from the start of milling until the fall of 1933, the mill crushed nearly 3,000,000 tons of ore, recovering about 0.167 ounce of gold per ton and leaving approximately 0.0325 ounce per ton in the tailing, indicating a recovery of about 80 percent. In the latter part of 1933, extensive repairs were made in the mill, flotation equipment for the treatment of plate tailings being added, among other things. Flotation, however, did not prove profitable, and after 14 months' operation was discontinued. Amalgamation is now the only recovery method used in the stamp mill, as in earlier days.

Ores

The ore in the Yellow Aster mine occurs mainly in the granites, although some deposits have been found in the schists. Virtually all ore mined has come from oxidized zones. The gold is free and rarely is accompanied by base metals, although a little pyrite, arsenopyrite, and scheelite are evident at times. Quartz and calcite are sparsely represented. Gold is sometimes observed, but the particles are generally quite small. The ore carries about 1 ounce of silver for each 3 ounces of gold.

Nearly all the ore-bearing granites are oxidized and soft, and gold usually lies in the small fractures and joint planes of the rock. When shattered or crushed, the rock breaks along planes of weakness, releasing the gold to the fines, while the harder, denser granite breaks into larger sizes, relatively barren of gold. So sharp a line of demarcation between ore and waste permits selective screening or sorting but presents difficulties in accurate sampling. It has been found that only by exercising the greatest care and by taking many and large samples can any reasonably close estimate of values be obtained. This same difficulty pursues the control of milling from the determination of battery heads to cyanidation residues.

Metallurgically, the Yellow Aster ore offers no unusual problems. The gold being free and there being practically no base-metal sulphides or earthy metal compounds deleterious to the methods of milling used, operating problems are mechanical rather than metallurgical. As in the case of nearly all desert places, water is the item of paramount importance and heaviest cost; hence all matters involving milling have their beginnings and endings in the economics of the water supply.

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MILLING

Milling at the Yellow Aster falls naturally into two divisions, one the stamp mill which operates on ore produced from the mine; the other the cyanide plant, which is designed primarily to recover gold from the 2,300,000 tons of impounded tailings but which also treats the current tails from the stamp mill. The stamp mill has a capacity at present (May 1936) of about 185 tons daily, and the cyanide plant handles about 1,100 tons daily, including tails from the mill. Water is exchanged between the two plants as the situation requires, the effort being to conserve every gallon possible. Thus, it will be seen that the two units are both complementary and supplementary to each other, although each can be and occasionally is operated independently. This paper will discuss each unit as a separate operation, bearing in mind that customarily the cyanide plant treats the tails from the stamp mill as they are produced.

Stamp Milling and Amalgamation

At the mine all rock hoisted is passed over grizzly bars spaced 3 inches apart. The oversize amounting to 30 percent is discarded as waste, while the undersize is delivered to a 60-ton-capacity crusher bin in 30-inch-gage, cradle-mounted, side-dump cars drawn by a 4-ton Plymouth gasoline locomotive. About 350 tons of rock are raised to the surface daily.

The ore in the crusher bin is reduced by a jaw crusher to a maximum size of 2-1/4-inches and conveyed by an 18-inch belt conveyor to a 4ø by 5-foot vibrating screen. Different-size screen openings are used, depending upon the areas from which ore is drawn. The oversize drops to a Hardinge constant-weight feeder, is automatically sampled as desired, and is conveyed to the waste dump by a 16-inch belt conveyor. The undersize goes to a similar constant-weight feeder, is likewise automatically sampled as required, and is transported and distributed to a 1,200-ton-capacity mill-storage bin by an 18-inch belt conveyor. At times the screens are removed, and all rock passing through the crusher is delivered to the mill bins. At present about 23 percent of the rock crushed is disposed of as waste, which, together with the 30 percent discarded in the mine, gives a total of 46 percent of the rock mined or drawn from chutes rejected as waste and 54 percent or 189 tons per day delivered to the mill as ore.

The mill consists of 50 stamps of 1,080 pounds each, arranged in batteries of 5 stamps, and each battery is fed by a Challenge feeder. The camshafts are driven by a line shaft extending the length of the mill and geared to a 150-horsepower electric motor. Thirty-mesh horizontal burr slot screens are used on the batteries. Each battery has a chuck plate, lip plate, splash plate, and a 5 by 14 foot apron plate, with riffles at the lower end. Back plates are occasionally used in some batteries with only moderate success. Mercury is fed to the batteries as well as to all plates, as required in the judgment of the millman. Apron plates are dressed daily.

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A general clean-up of plates and batteries is made monthly, the clean-up sands from inside the batteries are ground for 8 to 10 hours in an amalgamation barrel with an excess of mercury to effect amalgamation of gold and "floured quick." The "squeeze" is made in a screw press, and the resulting amalgam is treated in an oil-fired retort. The bullion sponge has an average fineness of 74 parts of gold and 200 parts of silver. The loss of mercury varies but averages about 0.9 ounce avoirdupois per ton of ore stamped. The flow sheet of the amalgamation mill is shown in figure 1.

Six and one-half tons of water is required per ton of ore crushed; part is fresh water and part is returned from the cyanide plant, depending upon the variation in the water-storage systems of the two plants. Plate tailings are thickened to some extent, depending upon water requirements and operating conditions at the cyanide plant, and flow down the hillside, joining the main stream of pulp from sluicing on its journey to the cyanide plant.

Excess water reclaimed from thickened tailings, containing a variable but appreciable tonnage of slimes, flows to two 40- by 8-foot tanks equipped with Dorr rakes. These tanks act as settlers, surge tanks, and storage for the sluicing operations of the cyanide plant, any excess in these tanks being repumped to the mill water storage tanks.

Plate tails carry 0.02 to 0.03 ounce of gold per ton.

CYANIDATION

The main purpose of the cyanide plant is to recover gold from the impounded stamp-mill tailings that have accumulated during the years the mill has run, treatment of currently produced tailings playing but a minor part in its design or operation. This has led to certain distinct features which might or might not be incorporated in a plant designed to handle mine ore. Reclamation of valuable metals or minerals from the waste products of previous operations, as conducted on a larger scale, is a business in itself, with certain important advantages as well as certain serious drawbacks. Among the advantages may be enumerated the following:

1. Relatively low cost of determining both tonnage and value of the waste material.

2. Relatively high degree of accuracy with which the determination may be made.

3. Relatively high degree of precision with which the outcome of a proposed venture may be predicted.

Among the drawbacks may be mentioned:

1. Relatively short life of operation.

2. Relatively narrow plant-cost limitations imposed by the first-mentioned disadvantage.

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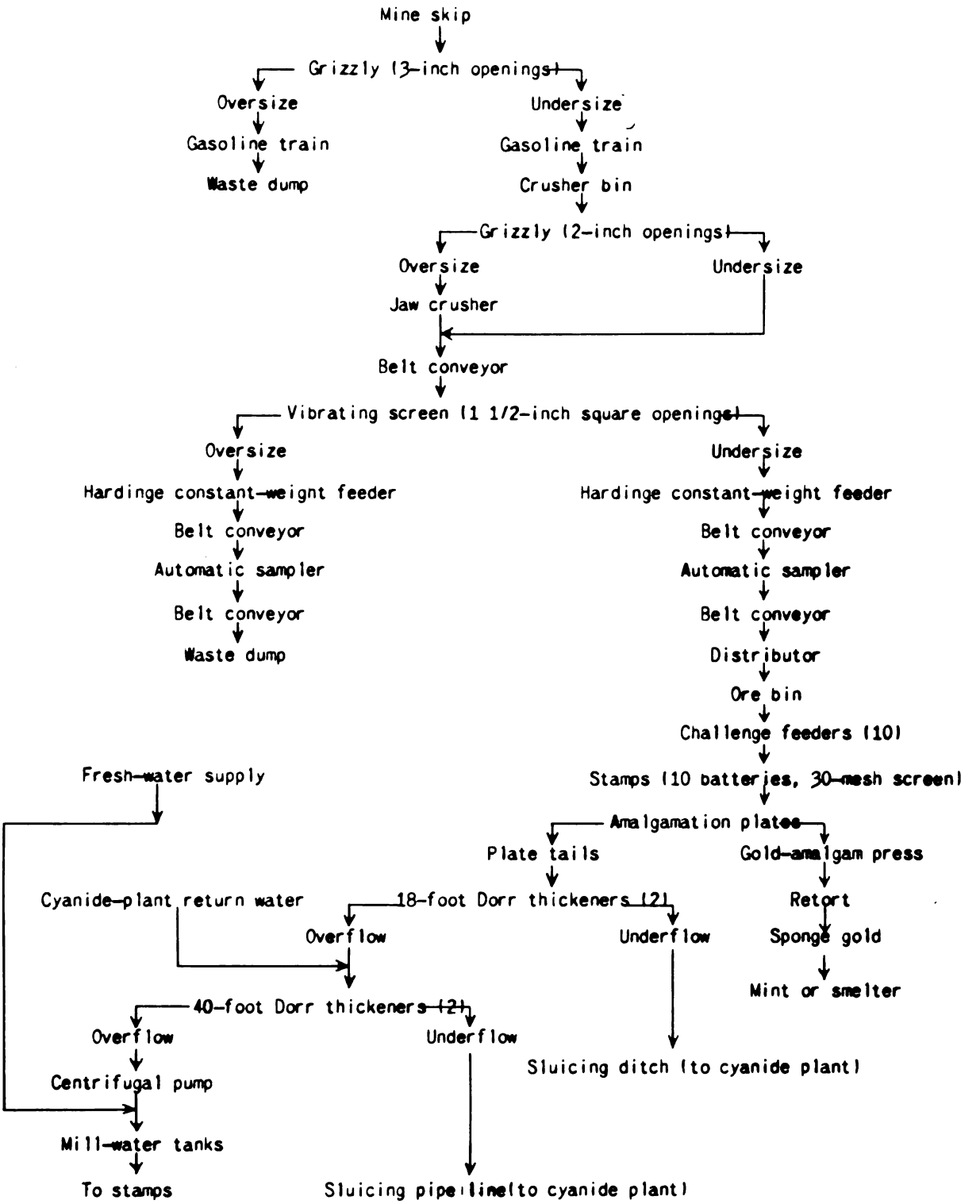

Figure 1.- Stamp-mill flow sheet, Yellow Aster mill.

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3. Temporary nature of all design, calling for closer supervision, makeshift expedients, and, generally speaking, higher maintenance costs.

Among other salient points may be included the thoroughness, with which the preliminary studies of the economic possibilities of the venture may be prosecuted at a low cost; the uncertainty of the influence of various kinds of debris (the nature of which cannot always be anticipated) in the deposit will have on recoveries, and as the material is already dressed, accurate decisions can be made as to size and type of equipment for the treatment plant.

Sampling

The higher price obtainable for gold induced the company, in the fall of 1934, to undertake a comprehensive sampling and testing campaign to ascertain the profit-making possibilities of reworking the Yellow Aster tailings. The firm of Hamilton, Beauchamp & Woodworth was employed to supervise the sampling (which was done by the local staff of the company), to conduct tests, design a plant, and supervise its construction.

Sampling was done with 2-inch ship-type augers and posthole augers; the latter were found to have a limited usefulness. Holes were drilled at the intersections of 200-foot coordinates mainly, although some irregularly spaced holes were put down before the work was fully organized. Holes were put down by hand, the auger being raised when driven the length of the worm. Five-foot sections of 3/4-inch pipe were added to the drill column as the hole was deepened; the 5-foot sections were removed and replaced, in the deeper holes, by 20- or 30-foot sections, as the column was raised or lowered. The unusual depth of the holes, ranging, in many cases, from 60 to over 100 feet, precluded the use of casing if the cost of the sampling was to be held within reasonable limits; all holes, with one or two exceptions, were drilled without casing. A short collar-piece, with hinged-steel flaps to support the weight of the drill column as raised and lowered, prevented top sands from falling into the holes. It is interesting to note that the net gain in depth for each pulling of the column was about the same as has been noted in other instances where casing was used. In the deeper holes it was necessary to pull the column with block and tackle attached to a light, portable tripod. In all, some 97 holes were drilled, representing a total footage of 2,950 feet, an average of over 30 feet per hole. Two crews of two to three men each required 45 days to put down the 97 holes.

Each hole was sampled in 5-foot sections, each section sample being assayed separately. A composite of each hole was also made by taking a unit weight of material from each section sample per foot of depth. A composite was finally made for each of the four blocks into which the entire deposit was divided, in the same manner that the hole composites were made, except that block composites were made from hole composites rather than from section samples. By this means it was possible to check calculated values against composites, both for holes and for blocks. Where composites failed to check section samples repeats were run and finally new composites made until the two values were within $0.08 of each other.

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At the same time a wet screen test, on a 200-mesh screen, was made on another portion of each section sample to gain advance information as to what might be expected in any particular area or elevation when mining.

Simultaneously, laboratory work was under way to determine recoveries as well as all the other data and information required to design a plant. This work was done on the composites of each of the four blocks, there appearing to be some difference in the sizings and other pulp characteristics in certain part of the dump. Testing followed conventional lines, no chemical problems appearing that suggested deviation from standard practice. The design and plans of the plant were carried forward with the laboratory work, since it was early decided what type of plant would give the best economic results. Erection of the plant was begun in April 1935, and it went into production September 1 of the same year, just 11 months after sampling was begun.

Mining Tailings

The Yellow Aster reclamation project consists of two parts. One involves moving the tailings from their original position to the plant and will be referred to as "mining." The actual treatment plant will be described separately.

The tailings are mined with water - the steep slopes of the hills on which they repose, the depth of the deposit, and the distance of the material from the plant indicated that sluicing would give the lowest cost, both initial and ultimate. Water for this purpose consists of the overflow from the dewatering thickeners at the plant, plus surplus water from the stamp mill, and such make-up water as is required to replace losses with plant residues.

Head for the nozzles is gained by utilizing the two 40- by 8-foot steel thickeners previously mentioned; these are on the point of a hill above the southeastern side of the deposit and slightly below the level of the stamp mill. Water is pumped from the plant to these storage tanks in two stages. The first stage is a 70-foot lift to three intermediate storage tanks having a capacity of 650 tons of water, which act as surge or swell tanks. Water is delivered to these tanks by a 4-inch centrifugal pump direct-connected to a 25-horsepower motor; the maximum capacity of the pump is 700 gallons per minute. Thence it is delivered to the two storage tanks by a 4-inch, double-suction, high-duty centrifugal pump, coupled directly to a 50-horsepower motor and having a capacity of 650 gallons per minute against a total head of 255 feet; the static head is 245 feet.

From the mining storage tanks a main water line leads down the hill roughly paralleling the east edge of the deposit. At regular intervals outlets are provided so that laterals may be led to any point of the dump where it is desired to mine. Enough 5-inch pipe, with slip-joints to facilitate rapid changes, is always on hand to permit taking off three or four laterals across the tailings as far as they can be mined from the location of the present main pipe line. Short sections of this 5-inch pipe have 2- or 3-inch outlets welded into them from which hose lines run to the nozzles. By this arrangement a large measure of flexibility is obtained, so that any point or points along

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the whole 2,000 feet of face may be mined as required on very short notice and with a minimum of labor. Pressure on the mining lines ranges from 30 pounds at the upper end to 85 pounds per square inch at the lowest point of the line.

The first step in the mining of the tailings is to break down the material by jets of water. For this purpose, 3/4- or 1-inch nozzles, or headers consisting of several 3/8-inch nozzles, are used. The latter device, mounted on a swivel at the termination of an extended pipe, has been found very effective in working down the very high banks without danger to men or equipment; it is used in one form or another almost exclusively against the sandier faces. The large single nozzles, in some instances, are mounted on tripods with Barco or Moran joints which provide the necessary movement of the nozzles while the hose supplies mobility to the unit. In other cases the hoses are merely strapped to tripods, the nozzle end overhanging enough to permit the required rotation. Water is carried to the nozzles by 2-inch or 2-1/2-inch rubber-covered hose.

The plan of mining is based upon using not only the power of the nozzles but also the undermining action of a flowing stream of water. Mining, therefore, is from the high side of the deposit downward, in two directions, in such a way that the water always bears into the banks as it flows along, thus increasing the water duty and decreasing the labor required. The terrain is very favorable to this sort of sluicing, the hills having, in all cases, enouch slope in two directions so that mining may be done without loss of efficiency at a number of places to the best advantage of the plant. The actual water used in moving tailings to the pump ranges from 2 to 4 tons per ton of dry solids, depending upon the material being moved. The hard, sun-baked slimes obviously reduce water effectiveness. Total water moved from the pump to the plant ranges from 2-1/2 to 5 tons per ton of dry solids.

The stream of pulp, as mined, flows down hill toward the lower end of the deposit; it collects from lateral cuts into a main ditch that traverses one block and terminates in the suction box of a 6-inch Wilfley sand pump, driven at 1,285 r.p.m. by a 60-horsepower motor through V belts. A plate punched with 1-1/8-inch holes covers part of the suction box, directly ahead of which is a vertical grizzly 8 feet long, with 1-3/4-inch openings. A water spray that extends full length of the grizzly accelerates the movement of coarser sand through the openings. Wing boards extend out about 50 feet from each end of the grizzly at an angle of 60°. The aim of this lay-out is to prevent pebbles large enough to clog the runner from getting into the pump and yet to provide a sufficiently large and trouble-free arrangement at the pump so that minimum attention will be required. The principal foreign matter in the tailings is gravel, accompanied by a few roots and twigs. Water from the mining main is piped to the pump, both for dilution and for cleaning screens. The suction box is provided with a safety overflow and with electric contacts that actuate howlers at various points up the ditches so the hoseman may be made aware of trouble at the pump station.

The 6-inch Wilfley pump is connected, through a 10-foot-radius hose bend (6-inch diameter) to the beginning of a 2,140-foot pipe line of 6-inch redwood pipe designed for 150 pounds working pressure. This line is connected by

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another 10-foot-radius hose bend at the plant to a 6-inch, iron-pipe, vertical riser extending 25 feet upward to the bottom of a discharge box. It is interesting to note that although the static head against the pump is nearly 35 feet the difference in elevation of the two ends of the redwood line is only 6 feet, the discharge end being the lower. In traversing the 2,140 feet the redwood line drops at a uniform grade to minus 25 feet, with respect to the pump discharge keeper yoke, thence rising at a uniform grade to its termination.

To insure quick unloading of the line in case of power interruption, a quick opening 6-inch valve is placed in the line at its lowest point, about 900 feet from the terminal. Removable collars and saddles with 1-1/2-inch outlets were inserted in the line at strategic intervals so that, should the line become clogged, it could easily and quickly be broken and flushed. But little trouble has been experienced on this account, only three complete stoppages having been noted, all due to special causes unlikely to be experienced again during the life of the operation.

The pump was speeded at the beginning of operations to impart a velocity of about 7 feet per second to the pump, at a density of 25 percent solids, thus allowing for moving 1,100 tons of dry solids in 20 hours. Mining operations now being spread over such long distances, with varying slopes on the hills and along such long faces, it is virtually impossible to maintain a uniform density or rate of flow (which has no practical importance anyway); hence all that can be accurately vouchsafed about the performance of the pump is that pressures vary over 24-hour periods from 40 to 85 pounds, with a probable average of 65 pounds, as shown by the recording pressure gage. Power input into the motor obviously varies through a wide range but is estimated to average around 50 to 55 horsepower, with much higher peaks.

Close records are kept of pumping hours and tonnages delivered by the pump. After an average life for wearing parts had been determined, parts were changed at the end of the span of duty without attempting to obtain the last hour of service, it being considered cheaper to sacrifice a little iron than to suffer delays due to parts failing during the night when only a minimum crew is on hand. Moreover, by following this policy no lost time has ever been charged to damaging motor or pump due to pulp coming through the casing. The average duty of a pump casing is 35,000 tons, while runners and follower plates are replaced after handling 16,000 to 18,000 tons of solids.

One foreman and four other men do the mining, there being one hoseman for each 8-hour shift, with an extra man on day shift to assist the foreman in shifting pipe and attending to the other odds and ends in connection with the mining. Every effort is made to relieve hosemen of the necessity of moving pipes; the heavier nozzles and headers are so arranged, when possible, that the nightmen may confine their attention to moving dirt.

Floodlights with 750-watt globes are used for night mining; they are attached to long, flexible, heavy-duty cords and moved from place to place as needed. Outlets are provided at regular intervals along a power line which extends from the pump station to the head of the dump. Telephone communication

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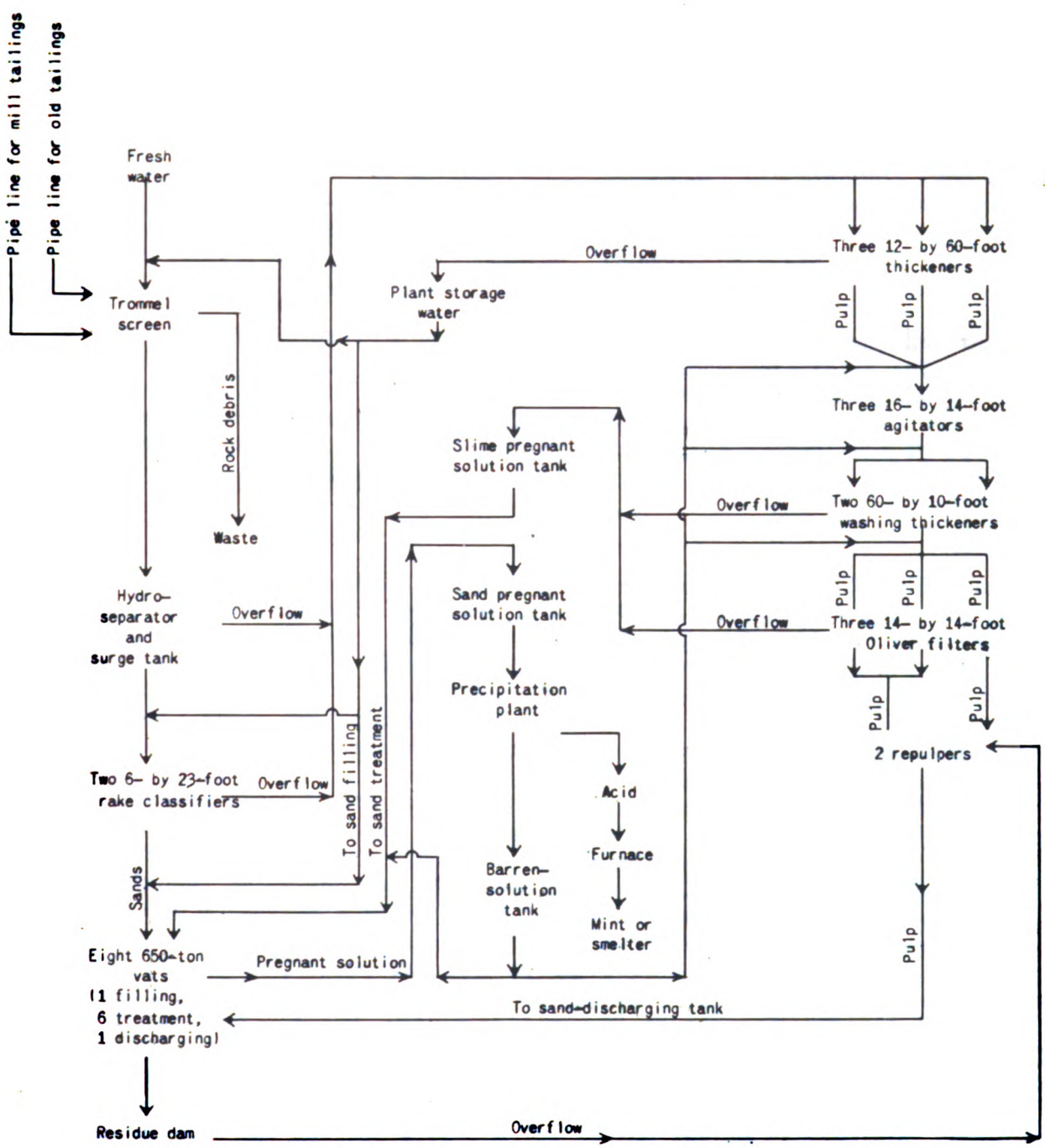

Figure 2.- Flow sheet, cyanide plant, Yellow Aster mill.

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is maintained between plant and mining operations by mine-type phones and sirens placed at the pump station and elsewhere along the faces. Recently it has been found that green lights, carried on a portable tripod and placed near where the hoseman is working, serve better than the howlers to attract his attention to a ringing phone.

Close cooperation between the mining operations and the treatment plant is absolutely essential and is demanded. Plant operators inform hosemen of the rate of feed and the condition of dewatering thickeners and transmit such other information as will assist the hoseman in controlling the water and mining. During the night shifts hosemen are required to report at regular intervals to the plant operators, more to promote further understanding than with the idea of checking up on any of the crew.

No estimate of the maximum capacity of the 6-inch Wilfley pump and the sluicing rate is possible. At no time since operations were begun has the mining operation been taxed to meet the requirements of the plant, and it is believed that only the limitations of the cyanide plant stand in the way of moving an average of 2,000 tons daily with the mining equipment now in use, and with no increase in man power.

Treatment

The flow sheet of the cyanide plant is shown in figure 2. The discharge of the 6-inch Wilfley pump is received in a 3- by 3- by 3-foot box placed over the end of the vertical riser pipe that terminates the pipe line. From there it passes by gravity into a 30-inch by 8-foot trommel screen with 1/2-inch perforations and set with a fall of 1-inch per foot; it revolves at 3 r.p.m. Small rocks and other debris discharge at the lower end into a chute and drop to a car below, which is wheeled away and dumped into the gulch when necessary.

Pulp passing through the screen is caught in a sloping apron or trough and carried to the feed well of the hydroseparator. This device consists of a 12- by 6-foot redwood tank, with an 8- by 8-inch peripheral launder set inside and 3 inches below the top of staves. A central, hollow-shaft gear driven at 3 r.p.m., extends to within 3 inches of the bottom of the tank; it has two rigid arms fastened to the shaft at about the level of the bottom of the peripheral launder. From the ends of these arms heavy chains are suspended and fastened again, at the bottom of the shaft, giving a slope of something like 3 inches to the foot. A rod, with a rubber ball on the end, extends from above the upper end of the hollow shaft through the bottom of the tank where the ball seats, from the outside, into a rubber disk with an orifice slightly smaller than the diameter of the ball. The rod and ball are balanced by levers and weights to provide any desired size opening in the bottom of the tank.

The function of the hydroseparator is to act as a surge tank for the two Dorr-type classifiers. In practice there is ordinarily little, if any, overflow from the hydroseparator. In the event, however, that sudden heavy overloads are delivered by the mining pump, it relieves the rake classifiers of part of the slimes. A typical screen analysis of the hydroseparator overflow is shown below:

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	Percent
Remaining on 100-mesh.....	6.4
Passing 100-remaining on 200-mesh.....	14.1
Passing 200-mesh.....	79.5
Solid in overflow.....	17.0

The two rake classifiers, 6- by 23-foot Model D Types, are operated at 22 strokes per minute on a slope of 2-1/16-inches per foot. Dilution of overflow is varied according to the rate of feed, areas being mined, and other factors influencing operations in the slime section. It may be controlled automatically, when desired, by the use of density bottles operating butterfly valves on the water lines. Baffles, spaced 12 inches from the overflow end of the pools and penetrating about 8 inches under the surface, are used when it is desired to overflow more coarse sand.

Dilution water is supplied from two sources: (1) A 4-inch centrifugal pump direct-connected to a 7-1/2-horsepower motor, which returns water from the dewatering thickeners; (2) the 4-inch Wilfley pump which normally returns sand-vat overflow water to the dewatering thickeners. The discharge from the Wilfley pump is deflected for use as dilution water only when additional dilution is required.

	Sands, percent	Slimes, percent
Remaining on 48-mesh.....	32.0	Tr.
Minus 48., plus 65-	43.4	2.5
Minus 65., plus 100-	14.7	20.9
Minus 100., plus 200-	5.5	12.9
Passing 200-	4.4	63.7
	100.0	100.0

No screen analyses of the plant feed are available owing to the difficulty of obtaining accurate or representative samples.

The classification units are more than ample for normal loads, although they were momentarily taxed severely by surges, these latter being inevitable under the mining conditions, as previously explained. A measure of flexibility, however, is always present, permitting variations in the separation of sand and slime that in the judgment of the superintendent are deemed advisable or in a lesser degree, of the operator on shift. This same flexibility in units, both as to capacity and performance, is striven for throughout the plant, experience in reworking old tailings having indicated this as an important requirement for the greatest measure of success.

Sand Plant

Sands from the rake discharge of the classifiers are washed by water down a main launder running between two rows of 45- by 12-foot redwood tanks, there being four vats to each row. Water is delivered to the head end of the launder by the same pump that provides dilution for the classifiers. Sand leaching tanks have conventionally designed false bottoms covered by cocoa matting, with

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12-ounce duck superimposed on the matting. The slope of the false bottoms is 1 inch per foot from outside to center. Nominal capacity of vats is 650 tons of dry sand each.

The sands are charged into the vats under water. They are distributed by power-driven revolving launders that extend across the vats to within 15 inches of the perimeters. Holes bored in the sides of the launders, together with small vertical deflecting strips, insure fairly uniform distribution of the sand while charging. Excess water containing some fine sands and colloids overflows the vats through a movable pipe held to within 6 inches of the top of the staves; this water discharges into the suction of the 4-inch Wilfley pump referred to above. This pump is direct-connected to a 20-horsepower motor and delivers upwards of 375 g.p.m. to the thickeners against a static head of 35 feet. No parts have yet worn out in this pump.

Vats are filled in rotation, and under normal operating conditions one tank is filled every 22 to 26 hours. When filled the feed is switched to the next tank to be charged; the drain valve of the filled tank is opened, the water draining to the 4-inch Wilfley vat overflow pump described above. At the end of 12 hours cyanide solution is flooded onto the top of the charge in the tank, and 2 hours later the drainings from the tank are turned into the solution circuit.

Treatment is then continued until 24 hours before the time to discharge the tank. Gold solution from the slime section is used exclusively during this period, which averages about 4-1/2 to 5 days. Solutions are added on the top of the charges only, and the ordinary routine is to permit at least two long drain periods for aeration during the time of treatment. Approximately 200 tons of barren solution are used for washing during the last 24 hours, after which the charge is thoroughly drained, sampled, and discharged. All solutions from the sand tanks are collected in a common sump called the "Sand pregnant tank."

The tanks are emptied by washing the sand into a central discharge valve of the Oliver type. This is accomplished by pumping slime residues, together with some water returned from the pool on the residue dam, into the revolving launders. Residue lines extend the length of each row of vats. They are connected to the discharge-valve opening by means of a hopper bolted to the bottom of the tank; they terminate in the suction box of the 4-inch Wilfley residue pump. High-pressure water, as well as barren solution, is tapped into the residue lines to clear any clogs as well as to provide dilution in the lines when required. Tanks are emptied in 10 to 14 hours, as required; the limiting factor is the capacity of the 4-inch residue pump. Relatively little attention is required for discharging vats after the operation is once fairly begun.

Slime residues and returned water from the dam are delivered to the top of the tanks by a 4-inch Wilfley pump, direct-coupled to a 20-horsepower motor. No parts have been worn out in this pump.

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A walkway, raised about 6 feet above the level of the sand vats, is centrally located between the two lines of tanks; branches extend to the center of each tank. Solution and water pipes are carried on towers beside and above the walkways, with valves for each pair of tanks grouped together. Since the classifiers, dewatering thickener pumps, and all solution tanks are either operated or are visible from the level of this walkway it is easily accessible from all points, either on the ground, or from the filter building.

The sand-plant operator normally occupies the walkway. He is charged with responsibility for the rate of mining, the control of classification, and the control of water and solution flows. Furthermore, he exercises general supervision over the activities of the slime plant operator.

Slime Plant

Fines overflowing the classifiers, as well as overflow and drain water from the sand vats, join in one central launder and are distributed to three 60- by 12-foot redwood thickener tanks equipped with worm- and gear-driven Dorr-type rake mechanisms suspended from trusses. Worms are directly coupled to a speed reducer motor; the rakes revolve at the rate of 1/7 r.p.m. Red or green indicator lights are hung over each drive to apprise the operator of any abnormal condition of the rake loads. Each thickener is provided with overflow launders. The clarified water drains into an 8-inch line laid on the ground parallel to the line of tanks and connected to the suctions of the two 4-inch centrifugal water pumps previously described. Feed to the thickeners varies widely in density but averages between 8 and 10 percent solids. These variations in density do not seem to have affected the capacity of the thickeners.

The thickened pulp is elevated 2 feet above the top of the staves by No. 4 Gould-type diaphragm pumps. A separate pump, direct-coupled to a speed-reducer motor, serves each thickener. Underflow from the dewatering thickeners averages 50 percent solids by weight and has the screen analysis previously shown. Thickened pulp from the three thickeners joins in a junction box at the head of no. 1 agitator. Fresh cyanide, milk of lime, and barren solution are added to the pulp before it enters the agitator.

Agitation is done in three 16- by 14-foot redwood tanks connected in series and equipped with Dorr-type agitating mechanisms revolving 3 r.p.m. Air at 10 pounds per square inch pressure, at the receiver, is used in the air lifts. No measure of the volume of air used has yet been made. Agitators are belt-driven by a 3-horsepower motor through a jack-and-line shaft. Agitator density is maintained at about 45 percent solids. No ill effects have been proved to result from thickening the pulp to 50 percent solids or from decreasing the volume of air used. It is felt, however, that there is a freer circulation with a reduction in the tendency to "short-circuit" by increasing the rate of agitation and thinning the pulp. Sodium cyanide is added in a sufficient quantity to maintain a strength of around 0.016 percent KCN, equivalent, which is permitted to fall as it passes through the plant until the barren solution will show a strength of only 0.012 to 0.013 percent KCN equivalent.

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Barren solution is added to the pulp as it leaves the last agitator. The pulp flows by gravity from the third agitator to two 60- by 10-foot redwood washing thickeners equipped with the same mechanisms as the dewatering thickeners, except that the underflow from these units is drawn off by gravity to a 3-inch, solid-lined, open-runner, Krogh centrifugal sand pump direct-connected to a 5-horsepower motor.

The average density of the underflow from the washing thickeners is 52 percent solids. This condition is attained by keeping the feed density within the range of the compression stage of this pulp which laboratory tests indicated to be around 22 percent solids. Thus a density equal to or heavier than that obtained in the dewatering thickeners is maintained, although the washing thickeners have only two-thirds the area and 56 percent of the volume of the primaries. The overflow solution from the washing thickeners is pumped by a 2-inch centrifugal pump, directly connected to a 5-horsepower motor, to a storage tank called the "Slime pregnant-solution tank", for distribution to the sand charges.

The underflow from the last thickeners is delivered through a 4-inch pipe line to a power-operated feeder that distributes the pulp to three 14- by 14-foot Oliver filters fitted with oscillating-type paddles to keep the sands in suspension. Filter speed is 1/4 r.p.m., with a cycle designed to allow 1-1/2 minutes for caking, 1/2 minute for drying, 1-1/2 minutes for washing time and 1/2 minute for blowing and recovering. A vacuum of 21 to 23 inches is maintained at the automatic valves, and the cake is blown off with air at 5 pounds pressure at the valve. With the type feed now going to the filters it is possible to dry the cake to 27-percent moisture.

Four rows of spray headers per filter are installed, with 16 nozzles per header, but in practice only three headers are generally in use, every effort being made to prevent dilution of the pulp in the pans. Washing is done with barren solution of which 40 to 50 percent of the weight of cake is used, depending upon the nature of the pulp. The header nearest the blowing point is supplied with only enough barren solution to keep the cake wet as it is being discharged. Experience has shown that since there is but little "blowback" of solutions from the sections, the cloth should be kept slightly lubricated with liquid; otherwise it discharges poorly and requires setting the scraper blades dangerously close to the drum. Various weights of twill are used as covers for the filters, wound with No. 10 gage wire at 2-1/2-inch spacing. Cloths are changed every 70 to 75 days. The cloth is still sound at the end of this period, but blinding with salts precipitated from the water has decreased its filtering capacity enough to warrant renewal. Filters are scrubbed every 5 days with steel-wire brushes, using a 5-percent hydrochloric acid solution. The beneficial effect is by no means startling. Each filter is driven by a 3-horsepower speed reducer motor direct-coupled to the worm shaft.

The final residue cake drops from the filters into repulpers. A single machine serves one filter, whereas another serves two filters revolving in opposite directions. The repulpers are of the blade type, revolving at 70 r.p.m. and individually driven by 3-horsepower speed-reducer motors through

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pinion and gear. Water pumped from the residue dam dilutes the pulp in the repulpers to any desired density. The diluted pulp flows by gravity to the sand vat sluicing pump previously described.

Filtrate from the filters is collected in a receiver from which it is delivered to the slime pregnant solution tank by a 3-inch centrifugal pump direct-connected to a 10-horsepower motor. Vacuum for the filters is produced by a 20- by 16-inch two-cylinder Ingersoll-Rand Imperial-type compressor, which is protected against solution coming over into the cylinders by a moisture trap and automatic check valve.

Filters and thickeners were rated to handle 450 tons daily of a feed containing 1 percent plus 100-mesh and 85 percent minus 200-mesh. However, in certain blocks of the impounded tailings it was found that just as good a recovery was possible in the slime plant as in the sand section on all sizes passing 65-mesh. Therefore, when working these areas the proportion of coarser material going to the slime section is raised, making possible an increase to 500 to 525 tons passing through the slime plant without materially raising the undissolved gold content residue and actually lowering the dissolved content of the residues.

Residue Disposal

As previously stated, all residues are joined in the suction of a 4-inch Wilfley sand pump, driven by a 40-horsepower motor through V belts. The pump discharges through 1,400 feet of 4-inch slip joint pipe to a 5-inch header pipe of the same type, extending the length of the face of the dam - about 625 feet at this writing. The 5-inch header pipe is carried on low bents, and when the face has built up to the top of these they are pulled up and shifted inward. The pipe is connected, in about 100-foot sections by rubber hose and clamps, giving it flexibility and facilitating the movement inward.

At 20-foot intervals along the header pipe a 1-1/2-inch outlet is made, over which a saddle with a 1-1/2-inch nipple inserted is clamped. Light flexible hose, with an internal diameter large enough to slip over the nipples and a length of about 16 feet, distribute the residues along the face; about six to eight hose are in use at any one time, being moved from saddle to saddle as conditions require. It is necessary to maintain a dilution of the residues sufficient to permit at least a partial separation of the coarser sands from the fines to build a satisfactory face. The excess moisture and fines flow toward the low place farthest from the face, where a pool of water collects. This pool is kept large enough in area to permit reasonable clarification before repumping to the plant to use over for sluicing. Briefly, this phase of the residue disposal amounts merely to "closed-circuiting" the residue dam with sand-tank sluicing operations. The average density of the pulp going to the residue dam, when emptying vats, is about 30 percent solids, though much thicker at times. The slimes residues are discharged well behind the face of the dam, so that no excess water will accumulate on the dam. One man, working 8 hours daily, takes care of the residue disposal. He is assisted by the repair crew when the 5-inch header pipe is to be moved.

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It may be of interest to add here that, due to the method of discharging sand vats, the rate at which residues are pumped to the dam often reaches a peak of over 80 tons of solids per hour. This rate is continued for several hours. All the material is handled by the 4-inch Wilfley pump previously mentioned.

Solutions

Before going to precipitation, all solutions are passed through the sand charges if possible. The effluent from the sand charges is collected in a 150-ton-capacity storage tank and pumped by a 3-inch centrifugal pump direct-connected to a 10-horsepower motor, to a 100-ton-capacity "gold tank", flowing by gravity thence to the clarifier tank. A butterfly valve controls the level of solution in both the gold tank and clarifier tank while a howler gives warning to the sand-plant operator should the level of the sand pregnant-solution tank get dangerously high. By this method the operator is required to give attention to only one gold-solution tank, thus reducing the risk of spillage.

The gold solution is treated by the Merrill-Crowe simultaneous clarification and precipitation system. The Crowe deaerating tower is direct-connected through a manifold to 20 Butters-type clarifier leaves. The leaves are 8- by 5-foot, and the clarifying medium is muslin. A filtering aid is coated on the leaves after washing, about one or two leaves being washed per shift.

A submerged, 3-inch, vertical centrifugal pump, direct-coupled to a 7-1/2-horsepower motor, withdraws solution from the Crowe tower and delivers it to the precipitate collecting bags. Zinc dust is fed into the suction pipe by a Merrill belt feeder and cone, a small amount of the pump discharge providing the circulation in the cone.

Precipitate-collecting equipment consists of two units of 60 bags each. Bags are 5 inches in diameter and 36 inches long, and are tied on bosses which, having internal threads in the center, screw onto nipples in the manifolds. The impeller of the bag pump is of such a size that it will not develop more than 6 pounds per square inch pressure on the manifolds when running at its maximum capacity. Barren solution flows by gravity from the bag tanks to two barren storage tanks of a total capacity of 200 tons. Between 1,750 and 1,850 tons of solution are run through precipitation daily. The barren solution will show only a trace of gold, except on the day of the clean-up, when it will sometimes assay as high as $0.01.

Barren solution is delivered to all points where required, except to filter sprays, by a 3-inch centrifugal pump which is direct-coupled to a 20-horsepower motor. The pump can deliver a minimum of 250 g.p.m. to any point required without robbing other units that take delivery from its discharge. A 1-1/2-inch centrifugal pump, direct-connected to a 5-horsepower motor, supplies barren solution for the spray nozzles, developing 30 pounds per square inch pressure at the pump under normal conditions.

Cyanide strength is built up mainly at the agitators. Very simple equipment keeps a steady stream of about 20 percent NaCN feeding into the pump. Lime is

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maintained at about 0.004 percent strength throughout the plant, mainly to aid settling rather than as a protective reagent against cyanicides. There are practically no cyanicides in this ore.

Clean-Up and Refining

Bags are removed twice a month, at which times the pressure on the manifolds has generally built up to about 5 pounds per square inch. Spare bosses with clean bags tied to them are screwed into place while the bags that have been removed are draining. In this way, precipitation is interrupted for only about 1 hour. About 5 pounds of a filtering aid is fed into new bags by the bag pump, after which about the same amount of zinc dust is also added. Barren solution is circulated back to the clarifier tank for 1-1/2 hours after precipitation is started before being turned into the barren tanks.

The contents of the bags are emptied into a lead-lined tank equipped with a power-driven stirrer, the bags being turned inside out and lightly brushed to remove adhering particles. Finally, after the melt is finished, the clean-up man washes the bags with hot water in a barrel-type, household, electric washing machine, dries and examines them, and ties the good ones to the bosses for further use. The wash water is returned to the acid treatment tank. Destruction of bags is inconsequential.

Precipitate sludge is treated with sulphuric acid, with the acid feed so arranged that it can be measured and drawn without direct human contact. No attempt is made to determine the acid strength in the treatment tank, enough acid being added to dissolve the zinc as quickly as possible. The solid residue is washed several times with water after action has ceased. The settled liquor is decanted into a filtering tank connected to the main vacuum system. The filtering medium is a canvas cover laid over a grid, with cheap gray filter paper placed thereon and a high-grade, unbleached muslin superimposed over the whole. A filtering aid is first coated over the muslin before filtering begins.

The washed precipitates are finally sluiced onto the filter and dried there as much as possible whereupon the shooting, with the entire mass of sludge, is transferred to pans, removed to the refinery, and dried over night in an enclosed electric drier.

Melting is done in an oil-fired circular furnace of the stationary type, using No. 125 clay-lined, graphite crucibles. These are handled in and out of the furnace by tongs attached to a winch on a revolving boom. Typical fluxes include 12 percent crude niter, 3 percent manganese dioxide, 25 percent borax, 20 percent soda ash, 20 percent silica, and 1 percent calcium fluorspar, these percentages being referred to the weight of dry precipitates. The crucible is charged continuously as the melting proceeds until nearly full of a liquid mass, whereupon, when fusion ceases, slag is dipped from the pot and charging is resumed until the entire weight of precipitate and flux has been introduced. Pouring is done from a rack into rectangular molds, slag being caught and chilled in a large container under the rack. From 125 to 160 pounds of precipitate is melted in 5 to 7 hours. Bars assay about 740 fine in gold and 220 fine in silver. Slags from the melts, of an average value of about 30 cents per pound, are accumulated in large lots for shipment to the smelter.

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The entire routine of the clean-up and melt occupies one man 2 days with the assistance of one other man for about an hour removing and putting on bags. In addition, the clean-up man spends about 2 days per clean-up washing, drying, and tying bags, sampling slags, and doing the other minor jobs in connection with this operation, after which he is free for other duties.

Reagents

About 80 percent of all cyanide used is fed to the slime agitators. A small amount is added to the first wash of each sand charge merely to offset the dilution of solution with water; it is hoped thus to accelerate the initial rate of dissolution.

Lime is fed to the Dorr classifiers for the dual purposes of assisting in thickening slimes and providing a small amount of lime for the sand charges. A mechanical slaker is charged with one shift's feed, which is elevated to the classifier by a pump. The lime now used is delivered already crushed to 1/4-inch-mesh and is rather slow to slake, so that the final milk fed contains some unslaked particles. These are dragged over by the rakes and result in an intimate mixture of sand and lime going into the vats, which, combined with the slow rate of dissolution of these particles, tends to produce a rather ideal condition in the sand charges.

A small amount of lime is added to the agitators, being slaked in batches and fed occasionally by hand to a stirrer in which a small stream of barren solution flows. This lime is added mainly to keep up the alkalinity in the washing thickeners, although there is no positive proof that it is absolutely necessary.

Zinc dust is fed at the rate of 0.04 pound per ton of solution going through precipitation. About 1 pound of sodium plumbite is added daily to the clarifier tank. This compound is made at the plant by combining caustic soda and lead acetate and is used preferentially because at times the alkalinity of the gold solution approaches or reaches the neutral point, as indicated by phenol, under which conditions an alkaline lead salt is indicated.

The consumption of major supplies is shown in table 1.

Water

The water used in this plant contains a rather high proportion of magnesium and calcium sulphates, as well as smaller quantities of aluminum sulphates, and divers other salts. While none of these compounds has any known deleterious action chemically and the water itself (having a pH of 7.8) is ideally suited for certain mechanical effects, nevertheless precipitation of the magnesium hydrate by lime has proved troublesome all over this district. This is one reason for holding the alkalinity at a low point. Actually, since the entire water circuit carries an alkalinity of 0.004 percent CaO, the precipitation of the hydrates is effected mainly in the water-storage tanks and in the dewatering thickeners. The effect of this is to produce a "tight" cake on the

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filters and to blind the covers. Because it is precipitated in the mining circuit and dewatering thickeners none has yet been found in the cyanide solutions so that it has never interfered with precipitation in any way.

TABLE 1.- Consumption of major supplies per ton of dry solids treated

		Sand plant	Slime plant	Total
Cyanide	lb.	0.008	0.304	0.180
Lime	do.	1.14	4.70	2.70
Zinc dust	do.	-	-	.076
Power	kw.-hr.	4.5	7.5	5.8
Water	tons	.7	1.3	.98
Average life of filter cloth.....days				75
Average life of 6-inch sand-pump casing.....tons				35,000
Average life of 4-inch sand-pump casing..... do.				72,000
Average life of 6-inch sand-pump runner..... do.				16,000
Average life of 4-inch sand-pump runner..... do.				19,000

Water for use around the plant comes from the main water-storage tanks at the mine level and is under a pressure of 200 pounds at the plant, which has both its advantages and disadvantages. The total water used at the cyanide plant amounts to about 0.90 ton per ton of dry solids treated. Being the largest item on the cost sheet, every effort is made to keep the loss at a minimum, the whole plan of operation revolving, as previously mentioned, around the most advantageous and economic use of water. It is noteworthy that evaporation appears to account for about 15 to 20 percent of the total water loss, with unrecoverable seepage and spillage in the mining operations accounting for about 5 percent, while the balance is absorbed in the residue dam.

Control

As accurate sampling of the daily feed to the plant is practically impossible, due to the many varying conditions, the value of the heads coming into the plant is determined by sampling sands and slimes separately. To determine the head value of the slimes the discharge of the diaphragm pumps of the dewatering thickeners is sampled hourly by the operators. Sand heads are sampled by boring the charges with a ship's auger; a number of holes are bored in each charge, arranged in a pattern that will make each hole represent roughly an equal volume of the tank.

A sample of the washing thickener-underflow discharge is taken at regular intervals over the 24 hours to determine both the undissolved and dissolved gold content of this pulp, reduced to the value per ton dry solids. Samples of the filter cakes are taken regularly, and these are consolidated into one sample for determination of the total residue and the break-down into dissolved and undissolved content. No attempt is made to record the performance of individual filters. Sand-residue samples are taken in the same manner as the heads. No attempt is made to determine the dissolved-gold content of sand residues, but a sample of the drain from each charge is taken when the residue sample is bored.

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Solutions are sampled by a home-made drip sampler, which seems to work very effectively, permitting close regulation and requiring but little attention. Included in daily solution samples are the bag heads and tails, as well as washing-thickener overflow and filter-filtrate solutions.

It might be of interest to note that a grab sample of the overflow of the dewatering-thickener overflow is taken every day, experience having shown that this water contains a certain amount of dissolved gold. The explanation is that, due to the necessity of reclaiming all water possible, together with the likelihood, on this account, of the operators mistakenly turning cyanide solution into the water circuit, there is always a certain amount of cyanide in the water circuit. Although only at rare intervals can the "free cyanide" be determined by ordinary titration yet a distinct dissolving action is constantly noted, with the result that the water circuit carries $0.03 to $0.06 per ton of water in dissolved gold. Desultory laboratory work since this condition has been discovered has indicated that a substantial proportion of the gold in the Yellow Aster tailings is soluble in solutions so weak that no trace can be discovered by common plant-control determinations.

Bag heads and slime-plant and sand-plant solutions are tested daily for free and total cyanide, as well as for protective alkalinity, the method described by E. M. Hamilton in his Manual of Cyanidation being used to determine free cyanide and the solution strength of the plant being regulated by the so-called "free-cyanide" strength. Due to the relatively large intake of fresh water daily little difference is noted between free and total cyanide. The samples tested are the 24-hour samples, no "grabs" being titrated unless for some unusual reason.

Very little change is observed in the average daily strength of solutions, either in cyanide or lime, and these changes are due mainly to fluctuations in tonnage or density as different areas in the tailings are mined. Variations in the quantities of reagents added are infrequent and made only on order from the foreman. No titrations are required of operators.

In the effort to obtain consistent and reliable assays on which to base records and control, 6 to 8 assay tons of pulp are taken of each sample of solids, while no less than 17-1/2 assay tons of solution are taken for assay, using the Chiddy method.

Operators take density samples of each thickener's discharge at regular and frequent periods; these are recorded on a daily report form which also indicates the agitator density, slime-tonnage determination data, and other information pertaining to the slime-plant operation. Another daily form records quantities of reagents being used on each shift, tons of solution precipitated, lost-time data, bag pressures, and other miscellaneous operating information. The record of each sand charge is kept on another form, which shows the tank number and the charge number (the latter running consecutively from the beginning of operations), the date filled, dry tons in the charge, and time charging and discharging, together with a record of the number and kind of washes going onto the charges. The mining section turns in a daily report showing hours of mining in various blocks or sections, the number of

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nozzles used during the different shifts, and any delays or lost time. This report, together with the 24-hour pressure-recording gage chart, gives a fairly complete picture of the mining activities.

In the very nature of things the tonnage passing through a plant of this kind cannot be determined by ordinary means. Even attempts to survey areas mined off are subject to wide variations and have been found more unsatisfactory than calibrations of the pulp as it moves through the process. In this plant, sand-plant tonnages are determined by measuring the level of the sand below top of staves, measurements being taken 2 days after charges have been under treatment to allow them to pack. At this point, tests indicate a volume of about 23 cubic feet per ton of dry weight, which factor is used in the estimates.

Slime tonnages are determined by what might be called a time-density method. At frequent intervals a large container (approximately 3 cubic feet) is passed under the stream of residue pulp from the filter and the time is noted with an electric clock, equipped with stop-start buttons, to fill the container to overflow. The container is suspended from a scales which records the weight. With these three knowns - time, weight, and volume - the rate of flow at the time the test is taken may be easily calculated, and by taking a number of samples over the 24 hours and averaging them, a fairly satisfactory estimate of the slime tonnage is obtainable. As a matter of fact, the application of the two described methods of tonnage determination has resulted, over the entire 7 months' operation, in an accumulated surplus of bullion over the indicated extraction (heads minus tails times tonnage) of less than one-half of 1 percent.

Personnel

The operating crew consists of 18 or 19 men, including superintendent and general foreman, divided as follows:

Mining	1 foreman and 4 hosemen
Treatment	6 plant operators
Clean-up and utility	1
Residue disposal	1
Repairs, maintenance, and general	4
Supervision	2

The assaying for mine, stamp mill and cyanide plant is consolidated in one office and the charges prorated to the various departments according to the use made of the service.

Climatic conditions at Randsburg leave something to be desired, more especially in the winter and spring seasons when high winds and wide fluctuations in temperature succeed each other in rapid order. Temperatures as low as 15°F. are quite common, with minimums considerably below that, all of which would mean nothing in a well insulated and heated plant but which add to both the difficulties and discomforts of the crew working virtually without

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protection and under somewhat wet conditions, especially in the mining section. Every effort is made, therefore, to protect men as much as possible under the conditions imposed by the character of the operation; although freezing weather has prevailed for as long as 2 weeks continuously, no loss of tonnage or equipment has yet been chargeable to this condition, a tribute to the cooperation and efforts of the entire crew.

Costs

Table 2 summarizes the direct operating costs during a typical month; data in table 1 are for the same period.

TABLE 2.- Operating costs, dollar per ton treated

Classification	Labor	Power	Water	Supplies	General	Total
Mining.....	0.0231	0.0209	0.0125	0.0103	0.0004	0.0672
Treatment.....	.0425	.0306	.0641	.0587	.0015	.1974
Clarification and precipitation....	-	.0010	-	.0074	-	.0084
Refining.....	.0050	-	-	.0008	-	.0058
Assaying and experimentation..	-	-	-	-	.0072	.0072
Residue disposal..	.0055	.0089	.0069	.0070	.0004	.0287
General.....	-	-	-	-	.0146	.0146
Total.....	.0761	.0614	.0835	.0842	.0241	.3293

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